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DEPARTMENT OF COMMERCE -- BUREAU OF MINES

METHOD AND COST OF MINING ZINC AND LEAD AT MINE NO. 2, TRI-STATE DISTRICT, PICHER, OKLA.

BY

WM. F. NETZEBAND

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METHOD AND COST OF MINING ZINC AND LEAD AT MINE NO. 2 TRI-STATE DISTRICT, PICHER, OKLA. [1]

By Wm. F. Netzeband[2]

INTRODUCTION

The mode of ore occurrence and the methods and costs of mining at one of the zinc-lead mines in the Tri-State zinc and lead district, Oklahoma, are presented in this paper for the information of mine operators in other districts.

HISTORY

No. 2 mine property was acquired by lease in 1918 after some 30 holes had been drilled and a shaft had been sunk to the 305-foot level by the first lessees. After acquisition of the lease, the present operating company drilled a rich ore body on the 260-foot level and in 1920 started sinking two shafts. Production was begun late in 1921, and the mine has been operated more or less continuously to the present time.

The mill has a capacity of 400 tons in 10 hours, the usual length of the mill shift in the district.

GEOLOGY

The geology of Mine No. 2 is very similar to that of Mine No. 1[3] except that the Cherokee shale is thicker, ranging from 80 to 160 feet, and all the formations are correspondingly deeper, the Short Creek oolite horizon here being at a depth of about 300 feet. The oolite has not been found on this property, but the lime comes in at about the level at which the oolite should appear.

ORE DEPOSITS

The ore deposits of Mine No. 2 occur in the brecciated and bowlder ground above the Short Creek oolite. The main or 260-foot level is about 40 feet above the oolite horizon, and the lower or 300-foot level is immediately above it.

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1 The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be made: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"

2 One of the consulting engineers, U. S. Bureau of Mines.

3 Netzeband, Wm. F., Bureau of Mines Information Circular 6113, 1929, 11 pp.

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The ore deposition is controlled by a zone of shearing and fracturing, and the mineable areas are divided into two distinct runs which have a northwest trend with a lean or practically barren area dividing them. This characteristic has controlled the mining to a large extent, for it enables the mine superintendent to locate the large pillars in lean ore and leave only small pillars in the richer ore. Figure 1 shows the outlines of the ore body and the pillars.

The ore on the main level occurs disseminated throughout the jasperoid breccia as patches or lenses in the jasperoid, or cementing massive bowlders of jasperoid breccia and chert. The ore of the lower level is more of the bowlder type than that on the main level and occurs chiefly cementing the bowlders but also with some disseminated ore in the jasperoid breccia.

The gangue minerals are jasperoid breccia, chert, and dolomite. Calcite is found in minor quantities. Sphalerite is the principal ore mineral; galena is found only in minor quantities around the edges of the ore body. When galena is encountered in this ore body, the limit of the ore can be expected.

EXPLORATION AND ESTIMATION OF ORE RESERVES

The exploration and estimation of ore reserves at Mine No. 2 are similar in many respects to the methods used at Mine No. 1.[4]

After the mine was opened up and the general character of the ore deposit had been determined, drilling operations were concentrated along the trend of the shear zone. Later drilling has proved what appears to be a parallel shear zone or an offset of the zone now being mined.

The shear zone was very difficult to drill before the water was drained off, and in consequence the cost of drilling was high. The early drilling cost $1.50 per foot, and many of the holes had to be finished on company time at the rate of $25 per diem. The present price of drilling is $1 per foot. On this property 61,080 feet of drilling has been done.

EARLY MINING METHODS

The methods of mining have not changed at this mine; the ore is still loaded by hand into cans; the cans are trammed by mules to the shaft and then hoisted to the surface, dumped into the mill hopper without being detached from the rope, and returned immediately to the shaft bottom.

The mining method, the open stope system with pillar support, has remained the same, but the character of the ground requires the use of larger pillars than at most mines of the district.

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4 Netzeband, Wm. F., Bureau of Mines Information Circular 6113, 1929, 11 pp.

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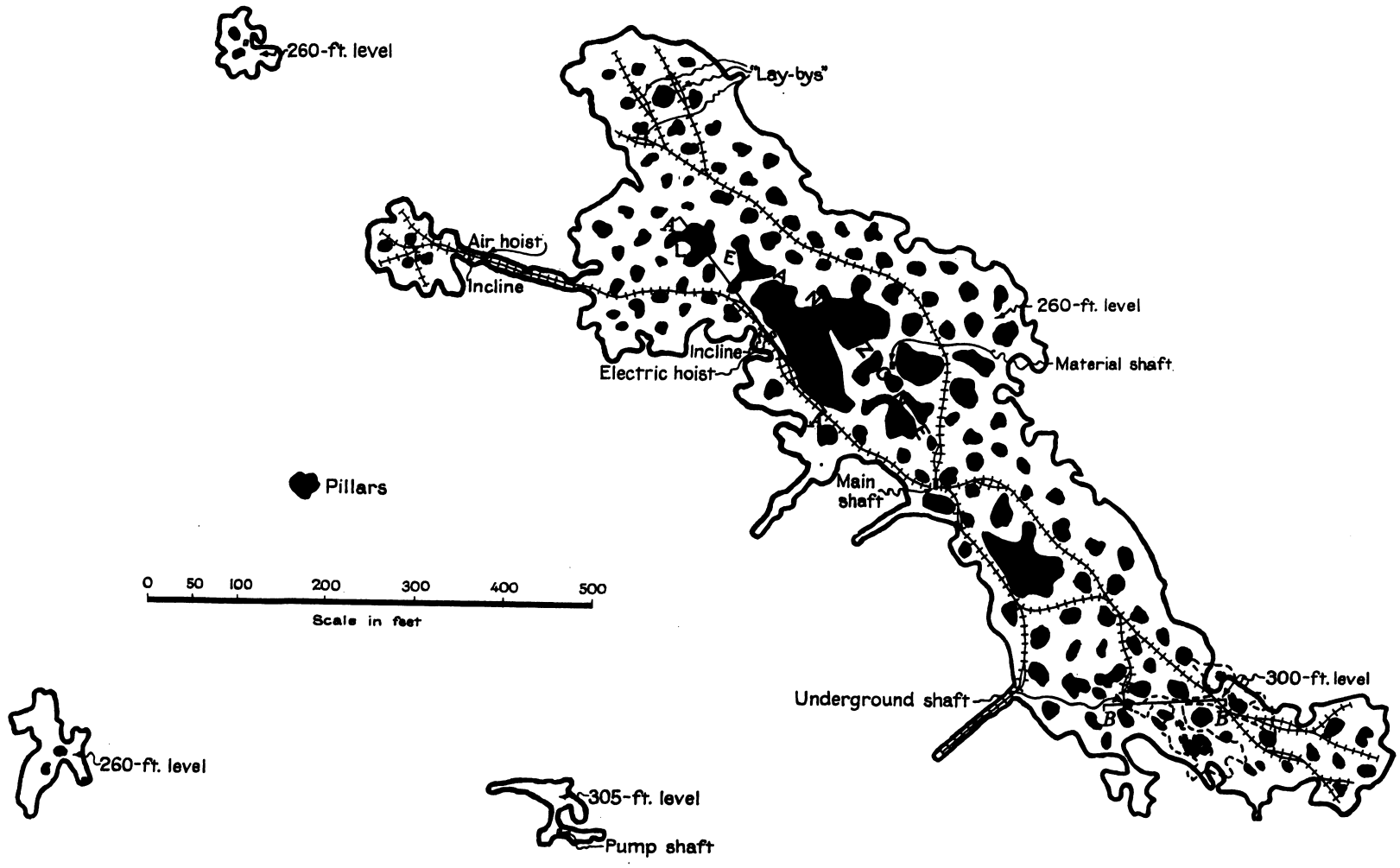

FIGURE I.—Plan of mine No. 2

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DEVELOPMENT SYSTEM

The first shaft was sunk by the original company to a depth of 305 feet. Considerable water was encountered, necessitating the installation of large pumps. When the present company took over the property, they drilled out a shallower and more extensive ore body upon which they sank two shafts. The original and deeper shaft has been retained for use as a pump shaft to drain the mine workings.

Main Shafts.- The main shaft was sunk as the mill shaft by the present company. The shaft is 5 by 11 feet in cross section. It was originally planned to install skips, but this has never been done. The shaft was sunk on contract, but no costs are available.

Auxiliary Shafts.- A field shaft was sunk at the time the mill shaft was put down, and the two shafts were connected by a drift on the 260-foot level. This field shaft is of the standard size, 5 by 7 feet in cross section, and is used for handling both men and materials; no men are handled at the mill shaft.

Two "field shafts" were sunk in 1926 on isolated ore bodies and were connected with the mill by surface trams. The sinking of these shafts was contracted for at the rate of $9 per foot in shale and $13 per foot in rock. The contractor furnished all labor and explosives and the company built the sinking derrick, installed the dump track, and furnished all equipment, material, and power.

The wages paid by the contractor were $5 per shift for the hoistman who also sharpened steel, $6 for the lead shaftman, and $5 for his helper. The contractor's costs were: Labor, 2 hoistmen $600; 4 shaftmen $1,320; labor sawing cribbing $40; explosives $410; liability insurance $104; a total of $2,474. The contractor was paid $2,980 for the completed shaft, which was 260 feet deep; 100 feet of the shaft was in shale.

The shaft was on an operating property so that air could be furnished by piping from the mine compressor. The sinking derrick was built to the dimensions of the standard hoisting derrick so that it could readily be converted into a standard derrick. The shaft was sunk dry. The following were the costs of the shaft to the company:

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16-foot sinking derrick	$ 160.00
12 by 14 foot "dog" house	45.00
Rent on 30-hp. boiler	60.00
Rent on steam hoist	60.00
Compressed air (estimated)	235.00
Fuel	90.00
Water for boiler	60.00
Cribbing and nails	443.00
Drill and equipment	335.00
Hoisting	110.00
Shaft and blacksmith tools	47.00
Oil	27.00
Sail and blower	144.00
1,600 feet of 2-inch pipe	346.00
Labor	80.00
	$2242.00
Shaft contract price	2980.00
Total cost	$5222.00
Total cost per foot	$ 20.08

The shaft is 5 by 7 feet in cross section and fourteen 5-foot holes constitutes a shaft round; a four-hole diamond cut is employed. The round is shown in Figure 2. The cut holes are drilled 5-1/2 feet in depth and all other holes are 5 feet deep. The side holes are drilled with a slight pitch towards each other. An average of 4 feet of shaft is broken per round, for which slightly more than a box of 30 per cent gelatin powder is used.

Most of the holes in the shale can be drilled with augers, but in the rock all holes have to be drilled with jackhammers. Four cut holes and four corner holes, using 40 sticks of 30 per cent gelatin powder, will advance the shaft 4-1/2 feet per round in shale. It is customary to break three rounds in the shale before cribbing. Where the flow of surface water is not strong, staggered cribbing is used, otherwise it is necessary to use solid cribbing. Care must be taken to put in enough lagging behind the cribbing, because the shale tends to slack upon exposure and thus to put an undue strain upon the cribbing. Figure 3 shows the method of supporting shafts in heavy ground. The foregoing information has been taken from a paper by S. S. Clarke[5], which was read before the Joplin-Miami Section of the American Institute of Mining and Metallurgical Engineers.

When water is encountered in sinking, pump seats are cut in the shaft at convenient intervals, and often as many as five or six of these seats are necessary. Sinking pumps suspended from chains or cable are seldom used in the district.

5 Clarke, S. S., "Shaft-sinking in the Tri-State district": Min. and Met., vol. 9, August, 1928, pp. 358-359.

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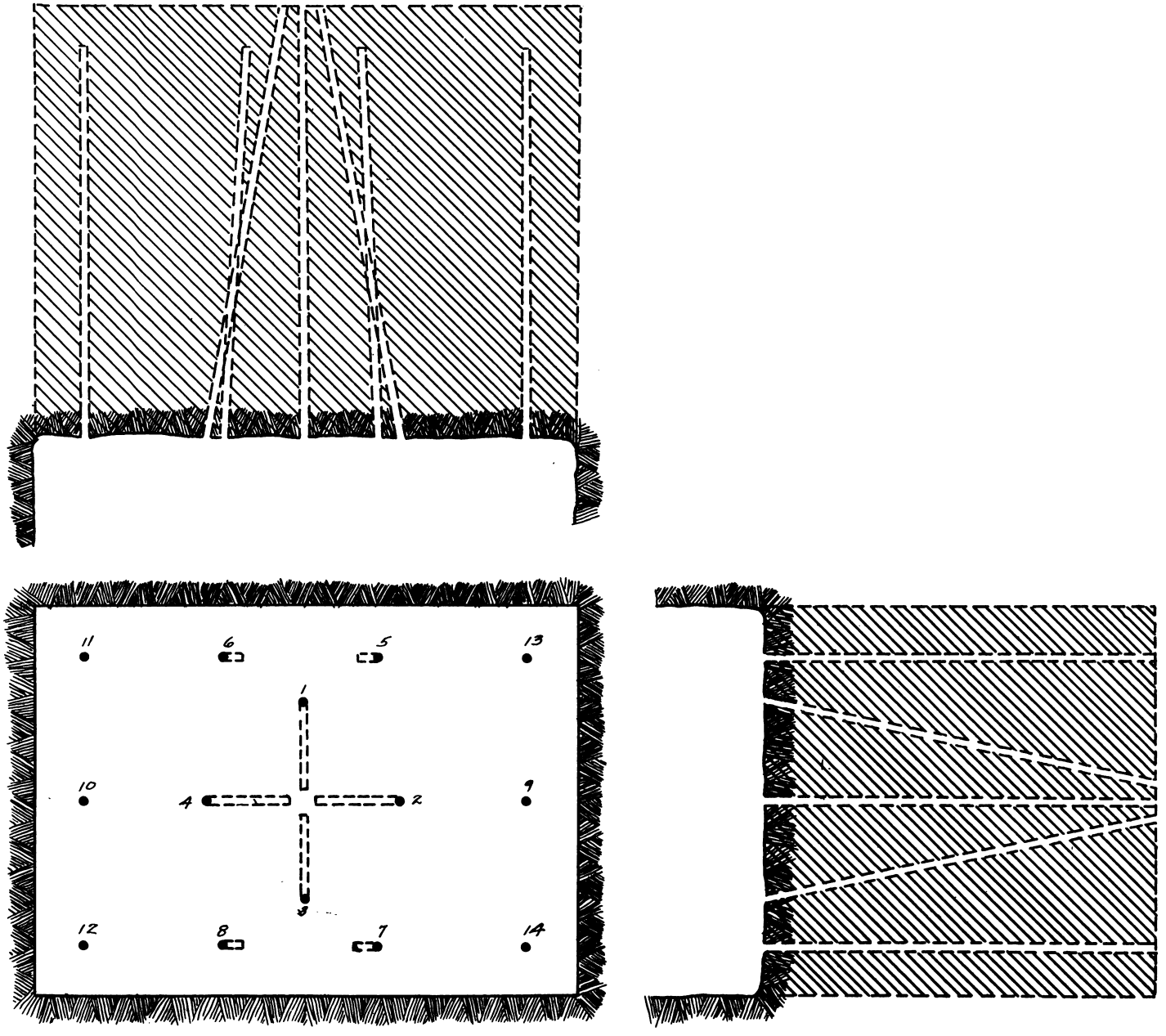

FIGURE 2.—Fourteen-hole shaft round

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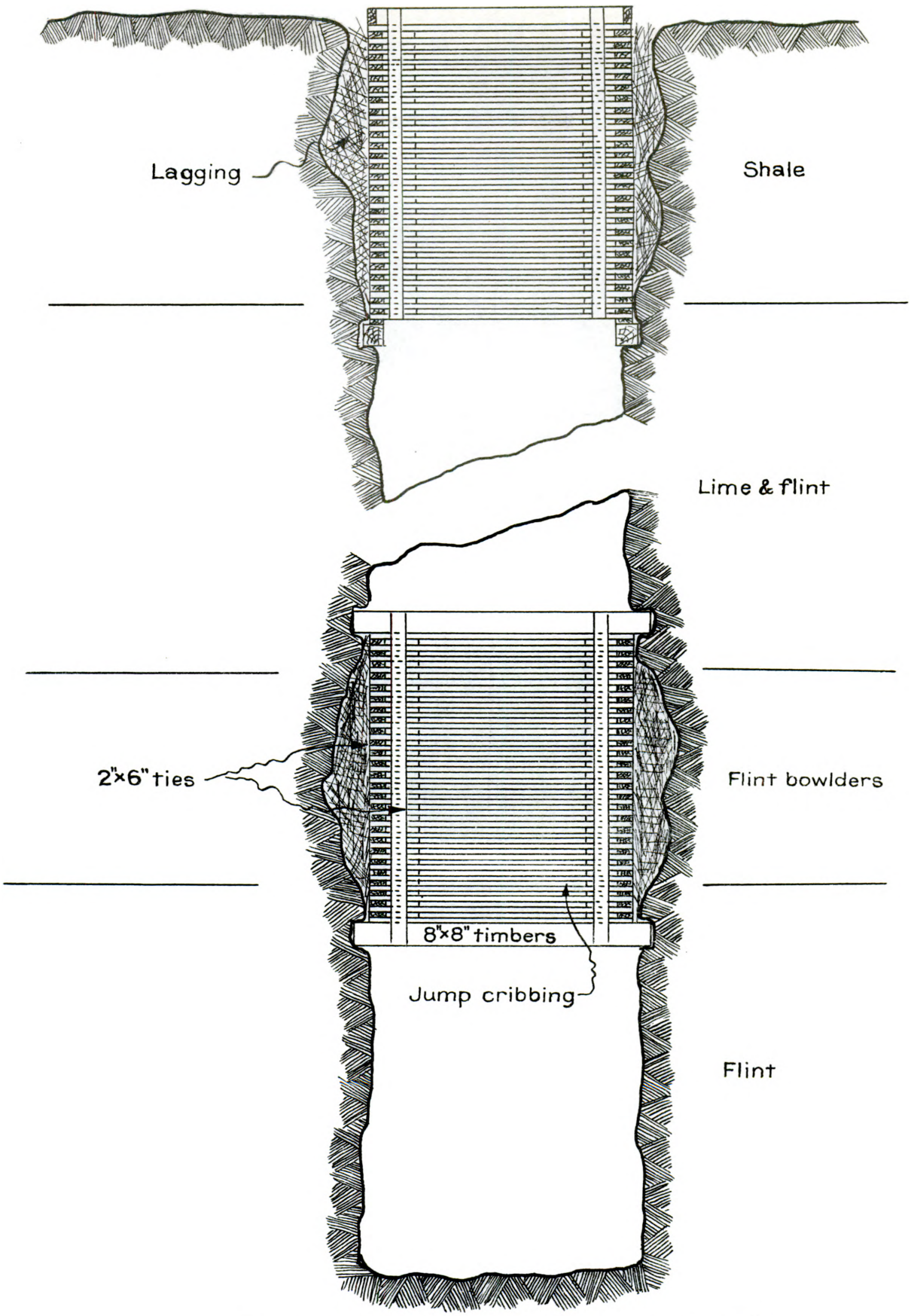

FIGURE 3.—Section of shaft, showing method of support in heavy ground

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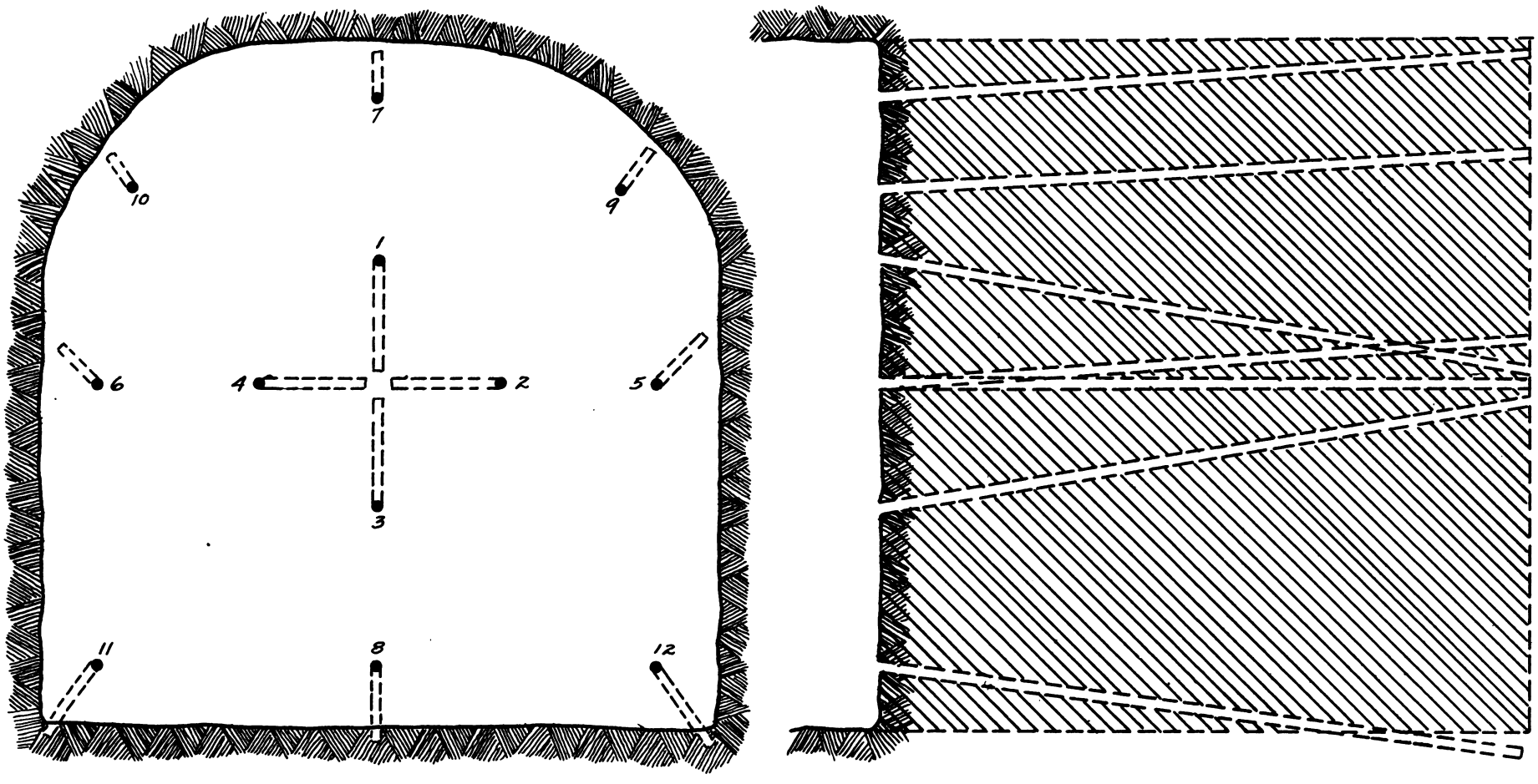

FIGURE 4.—Twelve-hole pull drift round for soft, spongy ground. For ordinary ground two cut holes are omitted

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An underground shaft or winze has been sunk from the main to the lower level through loose bowlder ground. It had to be cribbed and lagged very carefully. This shaft is the only connection between the main and the lower levels, and all ore, supplies, and men are handled through it.

"Pull" Drifts.- Prospect of "pull" drifts are driven 7 by 7 feet in cross section. Several drifts were contracted for at the rate of $9 per foot; the contractor furnished all labor, including the hoistman and the powder, and the company furnished the power and all equipment. No total costs are available for these drifts.

A drift was driven on company time at a cost of $6.50 per foot for labor and powder. The machineman was paid a bonus of 25 cents per shift, and the mucking was contracted for at the rate of $7 per round. The costs were as follows:

Drill labor	$ 1.46
Mucking labor	1.17
Hoisting	.79
Miscellaneous labor	.47
Explosives	2.61
Cost per foot	$ 6.50

A 10-hole round with two cut holes, two side holes, three roof holes, and three stope or bottom holes was used. About 24 sticks of 30 per cent gelatin powder were used in each hole, or about 2-1/4 boxes to the round. Usually 6 feet were broken per round.

For soft spongy ground a 12-hole round is used, as shown in Figure 4. The four-hole diamond cut, two side holes, three roof holes, and three stope or bottom holes constitute the round. The holes are drilled 8 feet deep; 7 feet are broken per round, using 2 boxes of 30 per cent gelatin powder. This ground is a soft, spongy, leached lime and flint and is known locally as "cottonrock." The drift is advanced whenever a machine is available from the regular mining operations, and the mucking is done on company time; no costs are available.

Raises.- No raises have been driven in this mine, for no mineable ore bodies have been found above the main level.

PRESENT MINING METHOD

The plan and sections of Mine No. 2 are shown in Figures 1, 5, and 5a. The headings are kept well in advance of the main stope except where the face is low, when no stope is carried; the entire face is advanced by the same method as the heading.

The ore breaks into large bowlders making it necessary to do a considerable amount of "bowlder popping" before loading into cans. All bowlders are drilled with jackhammers before blasting.

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For all drilling except the "bowlder popping" the heavy Leyner drill is used. The headings are drilled from a post, but the stopes are drilled from a tripod. For ordinary work 33 per cent ammonia powder is used, but for primers and "bowlder pops," 30 per cent gelatin powder is used. The cost of ammonia powder averaged $0.1315 per pound, and the gelatin powder $0.1300 per pound for the year 1928. The ratio of ammonia to gelatin is 5 to 1.

Underground Support.- The mine is supported almost entirely by pillars of ore from 15 to 70 feet in diameter, with an average of 30 feet. The larger pillars are left in the lean ore zone which runs through the center of the ore body, and the smaller pillars are left in the richer portions of the two ore runs. The pillars are spaced 40 to 60 feet, center to center, with an average of 50 feet

One section of the mine has a shale roof that becomes dangerous, requiring that the drift be cut down to a "pull" drift size and necessitating the use of several sets of caps and posts with lagging. This method of support is of minor importance. No mud sills are required, for the floor is hard.

The roof stands well if it has been properly arched and carefully trimmed as the faces are advanced. All mines in the district employ a "roof trimmer" whose sole duty is to examine all working faces every day and keep the walls and roof free from loose rock. Most of the trimming can be done from the top of the broken ore or from the heading floor, but for work too high to be reached in this manner the "trimmer" employs a ladder. Ladders are in 20-foot sections, and in very high headings three sections of ladder are often needed. The "roof trimmer's" tools comprise a short bar of 7/8-inch hexagon steel which has been pointed at one end and shaped like a nail bar at the other end. This bar is 6 to 10 feet long. For places that can not be reached with this, a longer bar, made usually of 1-inch pipe, is employed. The "roof trimmer" wears the ordinary carbide cap lamp, but usually a larger reflector replaces the one which comes with the lamp. The reflector from a Model T Ford car with the nickle plating removed makes an ideal lamp reflector, although it is a little heavy for continuous wear.

DRILLING AND BLASTING PRACTICE

Compressors.- Air is furnished by two belt-driven, two-stage compressors. The main compressor has a capacity of 1,050 cubic feet per minute, and the booster has a capacity of 950 cubic feet per minute. The pressure is maintained at 110 pounds at the compressor, which gives an effective pressure of 95 pounds at the machines.

Drills.- Several makes of drills are used but they are all of the heavy Leyner type. Some of the lighter drills can be run as one-man machines, but they are all used as two-man machines at this property. The drills are mounted on posts for heading and "pull" drift work and on tripods for drilling the stopes. All Leyner drills use 1 1/4-inch hollow round steel and the jackhammers use 1-inch hexagon hollow steel.

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The standard cross bit with an 18° taper is used. The gauges are dropped 1/4 inch for steel length changes up to 11 feet and then the gauges are dropped 1/8 inch for steel changes up to 20 feet. The starter bits are three feet long with a gauge of 2-3/4 inches. The steel lengths increase 2 feet for each change except the last one, which is 3 feet. The 20-foot steels have a gauge of 1-1/2 inches. The steel consumption is 0.008 pounds per ton of ore mined.

The blacksmith shop is located on the surface near the collar of the material shaft. All drill steels are bitted and shanked here on a power sharpener.

Blasting.- The blasting practice at Mine No. 2 is essentially the same as that used at Mine No. 1, but the character of the ground is different enough to warrant a different type of heading round and different loading of the "squib shots."

The heading rounds are drilled from a 7 1/2-foot post using 10-foot steels. A round consists of seven to nine holes. The nine-hole round is shown in Figure 6, which also shows the manner of drilling the stopes or benches. Two boxes of powder are used for each round, and about 10 feet are broken per round.

The faces are not so high in Mine No. 2 as at Mine No. 1. The high faces average only about 25 feet, so that most stope and splitter holes are drilled only 12 to 15 feet deep. The slope of the stope from the crest to the floor is maintained at about 45°. For a 25-foot face the collars of the splitter holes are about 10 feet behind the crest of the stope, and the collars of the stope holes are the same distance behind the collars of the splitter holes.

The splitter holes are three in number unless the distance between pillars has to be cut down because of unsafe ground, when the number is two. These are drilled in the same horizontal plane and have a rise of 1 inch to the foot so that they can be readily washed free of cuttings after squibbing.

The stope holes are always the same in number as the splitters and are placed practically vertically under the splitters. These holes are drilled downward with the same fall as the rise of the splitters. In order to prevent high bottom they are so located that the bottom of the hole will be below the grade of the drift.

Each splitter and stope hole is squibbed or chambered twice before the breaking charge is loaded. Five sticks of powder are used the first time. The hole is then thoroughly washed, and 24 hours later it is shot with 15 sticks of powder. The hole is now ready for the breaking charge. The breaking charge takes from one and a half to two boxes of 33 per cent ammonia powder for the stope holes and one to one and a half boxes for the splitter holes. The pockets are always loaded to capacity, and two primers are used to insure against a misfire.

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All primers are made up with 30 per cent gelatin powder, using a No. 6 cap. For the rest of the blasting, except in the "pull" drifts, 33 per cent ammonia powder is used. The cap is placed in the center of the stick of powder and the entire primer is placed in a safety tube provided for the purpose. Waterproofed fuse with a speed of 2 feet per minute is employed. All charges are stemmed, using clay cartridges made from clay found on one of the company's properties. The clay cartridges cost about $0.01 each.

The superintendent endeavors to maintain his tonnage at 100 cans or about 60 tons per machine shift. This includes machines on heading and stope work. Many of the faces are low, the average height of face in this mine being 14 feet.

LOADING AND TRAMMING

All loading is by hand shovelling into cans set on low trucks. The cans are 32 inches in diameter by 30 inches deep, and have a capacity of 0.6 tons of ore. The track gauge is 18 inches.

All tramming from the "lay-bys" near the working faces to the hoisting shaft is done by mules. There are four mules in the mine, three in daily use and one in reserve for emergencies. On an even grade a mule can pull 12 loaded cans. Figure 1 shows the haulage plan and lengths of hauls.

The northwest part of the mine is 10 feet lower than the main level, and hoists are necessary to haul the loaded cans up the inclines. On the main haulageway a geared, belt-driven hoist is used (fig. 5). A 10-hp. motor furnishes the power. This hoist is capable of handling six loaded cans per trip. Two short inclines from the working faces are operated by small air hoists or "tuggers." These hoists can handle only one loaded can at a time.

The ore from the lower level is brought to the main level by a 52-hp. geared electric hoist.

PUMPING

Ordinarily no pumping is now necessary, since adjoining mines working at lower levels keep the water level below the present workings. In the spring of the year when the rains cause a heavy flow of surface water, the pump at the deep pump shaft is used.

During the summer months the pumps are run to supply mill water for several of the company's mines, which are connected with the pump shaft by a 10-inch pipe line.

The pumps are electrically driven and have a capacity of 450 gallons per minute.

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LABOR EFFICIENCY, 1928

Record of labor performed at Mine No. 2 for nine months of 1928:

	Total Shifts	Tons per Shift
Trammers	1,032	84.84
Drill runners and helpers	2,924	29.94
Muckers	3,784	23.14
Miscellaneous Labor	2,064	42.42
Total underground operations	9,804	8.93

Ore hoisted during 1928 amounted to 87,563 tons.

PERCENTAGE OF EXTRACTION

At present 25 per cent of the area mined is left in pillars. It is expected that some of these pillars can be recovered, so that the total extraction will be about 80 per cent. Many of the pillars will be left because they are too lean to pay the recovery cost, but the greater number must be left to protect the surface because of the character of the ground.

WAGE AND CONTRACT SYSTEM

All labor, except muching, is based on an eight-hour day. Muckers are paid on contract, 11 cents per can of 0.6 tons on the main level and 14 cents per can on the lower level. The bonus is paid to men employed on the lower level because the ventilation is poorer and they can not compete on an even basis with the men on the main level. The following wage scale was in effect during 1928 when zinc ore prices stayed at $40 or under for prime western ore.

Machine runners	$ 4.25
Machine helpers	3.75
Trammers and drivers	3.50
Hoistmen	4.75
Powdermen	4.50
Roof trimmers	4.00
Screenmen	3.50

This wage scale is based on $40 zinc ore. If the price goes to $45 and stays there for one week, all wages are automatically raised 25 cents per shift, and muckers are raised 1/2 cent per can. In the same manner, for every $5 raise in the price of zinc ore above $45 the wages are raised at the same rate, but for every drop of $5 the wages are reduced. Wages are not reduced when the ore price drops below $40, for this is the base price for the wage scale. A week is always allowed between the wage changes to make sure that the market will not fluctuate above or below the critical price.

An average experienced mucker will load 40 or more cans in an eight-hour shift. There are several men at the mine who will average 100 cans per shift, but they are exceptional men and are usually given the best working places.

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VENTILATION

The ventilation is natural, except in one place where an ore body is being opened up at the end of a long "pull" drift. A blower has been installed on the surface and air is forced down to this area through a churn-drill hole.

A large blower has been installed at the man shaft to change the draft in the winter time in order to prevent the formation of ice.

MINING COSTS

Below is the total cost of delivering a ton of ore into the mill hopper at Mine No. 2:

Cost of delivering 1 ton of ore into the mill hopper

	Mining	General underground	Surface expense	Total
Labor	0.392	0.084	0.055	0.531
Supervision	0.045	- -	- -	0.045
Compressed air, etc.	0.114	- -	- -	0.114
Power cost	- -	0.025	0.051	0.076
Explosives	0.157	- -	- -	0.157
Other supplies	- -	0.056	0.014	0.070
Total	0.708	0.165	0.120	0.993

The surface expense, which is directly applicable to underground operations, includes the hoist and screen room and the surface tram from the field shafts to the mill hopper; the incline tram into the mill hopper is also included. Ore was mined from field shafts during only a small part of the year; the major part of the ore was hoisted at the mill shaft and dumped directly into the mill hopper.

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SUMMARY OF COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

Name or number of mine: No. 2, Tri-State district — Year: 1928, 9 months' operation

Tons of ore mined and hoisted 87,563

Mining method: Open stope with pillar support.

(A) Labor (Man hours per ton)

Breaking (drilling and blasting)	0.297
Mucking	0.345
Haulage and hoisting	0.125
Supervising	0.031
General	0.078
Total underground labor	0.876
Average tons per man-shift underground	8.93
Labor, percentage of total cost	58.00

(B) Power and Supplies

Explosives, 33 per cent ammonia (lbs. per ton)	1.265
Total power (kw. hrs. per ton)	5.75
(1) Air compression	2.51
(2) Hoisting	2.42
(3) Pumping	0.82
Other supplies in percentage of total supplies and power	16.78
Supplies and power, percentage of total cost	42.00

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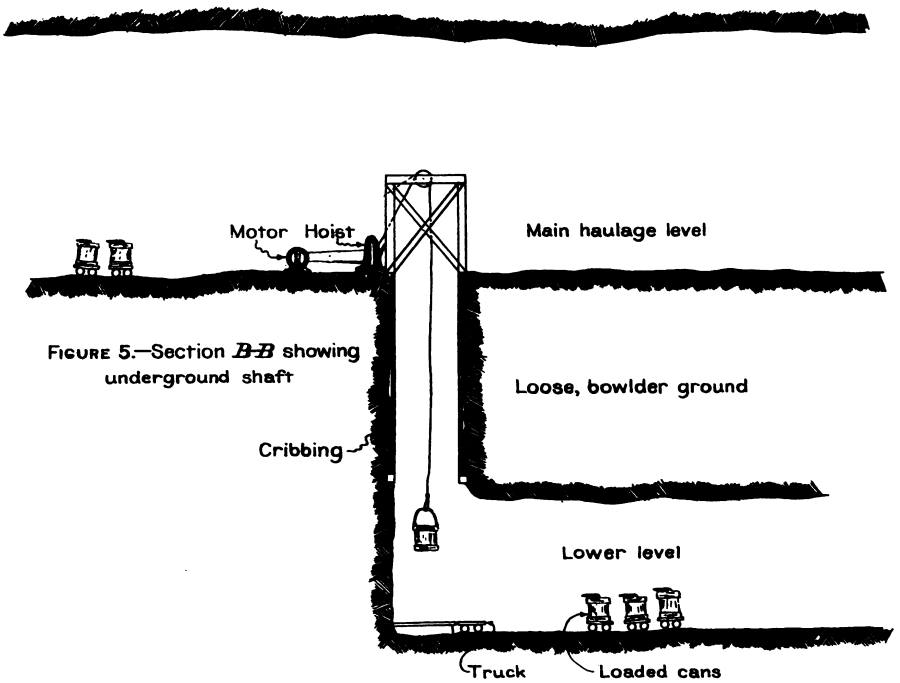

FIGURE 5.—Section *B-B* showing underground shaft

FIGURE 5A.—Section *A-A* showing incline-hoist layout

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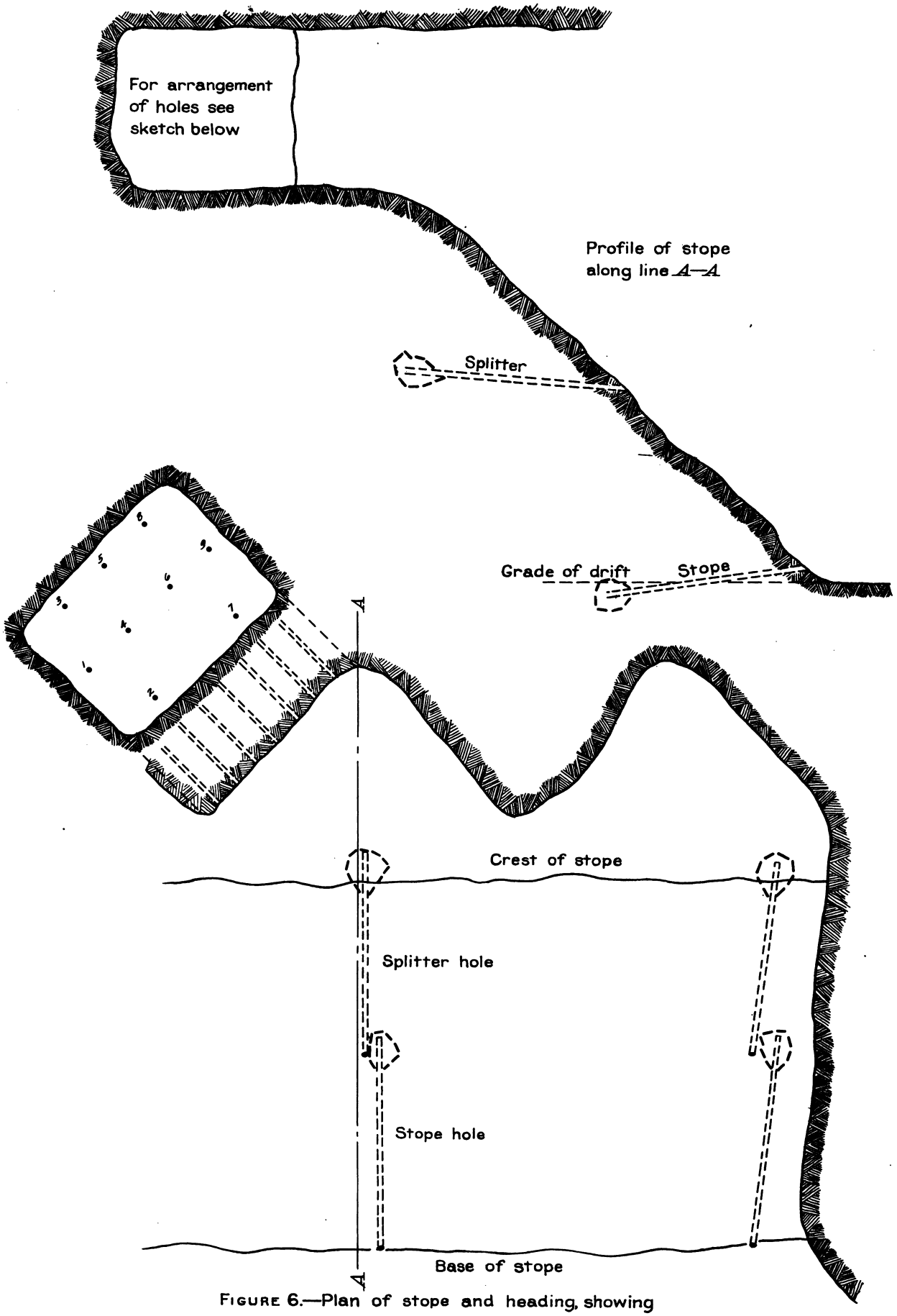

FIGURE 6.—Plan of stope and heading, showing arrangement of holes

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